# A method for predicting the specific energy requirement of comminution circuits and assessing their energy utilisation efficiency

S.Morrell

*SMCC Pty Ltd, 29 Camborne Place, Chapel Hill, Qld. 4069 Australia*

**Abstract**

This paper presents a new approach for determining the specific energy requirement of tumbling mill (grinding) circuits. It is used to demonstrate that there is no significant difference between the energy utilisation efficiency of any of the grinding circuits of the plants studied. Laboratory test results are also presented which indicate that classifier efficiency and recycle load in closed ball mill circuits can have a significant difference on the apparent energy utilisation efficiency. The experimental data indicate that this is achieved through changes to the gradient of the size distribution of the final product.

## 1 Introduction

The currently most popularly accepted beliefs on the relative efficiency of different grinding circuits have been strongly influenced by views expressed in the 1980's and are encapsulated in the following quotes:

"...(compared) with conventional grinding.....power in autogenous and semi-autogenous mills is transmitted less efficiently. Autogenous mills are usually less efficient than SAG mills" (Barratt and Allan, 1986)

"...SAG mills are inherently less efficient than rod mills" (Knight et al, 1989)

"....over conventional circuits......one disadvantage is generally the increased power costs for semi-autogenous circuits..." (Thomas, 1989)

Two sentiments are reflected in these quotes, viz. that different circuits in general have different energy efficiencies and that conventional circuits, which at the time the quotes were made related to crush-rod-ball and crush-ball circuits, were more efficient than AG and SAG circuits. Interestingly, data to support these views are almost completely lacking in the published literature. Bond's landmark papers of 1961 contain equations which explicitly include efficiency factors and their application results in different circuit configurations being predicted to require different specific energies to reach the same final grind, ie it is implied that different circuits will have different efficiencies. However, the Bond approach is predicated on the assumption that size reduction follows a particular relationship as described by the well known "third theory" equation (Bond 1952):

$$W = W_i\left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right) \qquad (1)$$

where

| | | |
|---|---|---|
| W | = | Specific energy |
| Wi | = | Work index |
| P | = | 80% passing size for the product |
| F | = | 80% passing size for the feed |

On the basis of this equation, the higher that the specific energy of a circuit is compared to what this equation predicts, the less efficient it is assumed to be. Due to the widespread acceptance and use of Bond's equations over the last 50 years comparisons that have been made of different circuit designs have invariably relied on his approach, particularly the use of so-called "operating work indices" (OWi). The operating work index associated with Bond's equation is written as:

$$OW_i = W \Big/ 10\left(\frac{1}{\sqrt{P}} - \frac{1}{\sqrt{F}}\right) \qquad (2)$$

In theory the operating work index can be used to compare different circuits which have different feed and product sizes. Providing the ore is the same, higher operating work indices imply less efficient circuits. Different circuits treating different ores in theory can also be compared by using the ratio of the operating work index to the laboratory work index. Hence a circuit with an operating work index of say 23 kWh/t and a laboratory work index of 20 kWh/t will have a ratio of 1.15. This ratio is often referred to as the "$f_{sag}$" of the milling circuit. If a different circuit treating an ore with a laboratory work index of say 16 kWh/t has an operating work index of 16.8 kWh/t its ratio will be 1.05 and it is therefore considered to be more energy efficient than the former and therefore has a lower "$f_{sag}$". However, the conclusion that the latter circuit is more energy efficient than the former will only be true if the performance of the 2 circuits obeys equation 1.

It is noteworthy that if the $f_{sag}$ ratio described above is determined for a wide range of operating plants and compared with the DWi value (a measure of ore harness from an AG/SAG/Crushing and HPGR perspective (Morrell, 2004a)) the trend shown in Figure 1 is obtained. The result suggests that the apparent efficiency of a grinding circuit as described using Bond's equations is related to the hardness of the ore, as $f_{sag}$ increases with increasing DWi. The implication of this result is that if an approach based on Bond's equations is adopted to determine the specific energy of a grinding circuit the application of a factor such as $f_{sag}$ may be appropriate. However, it is emphasised that this does not mean that the energy efficiency of a grinding circuit decreases as the AG/SAG ore hardness increases, merely that the Bond equations progressively depart from reality and may need a correspondingly larger correction factor ($f_{sag}$) to obtain accurate results.

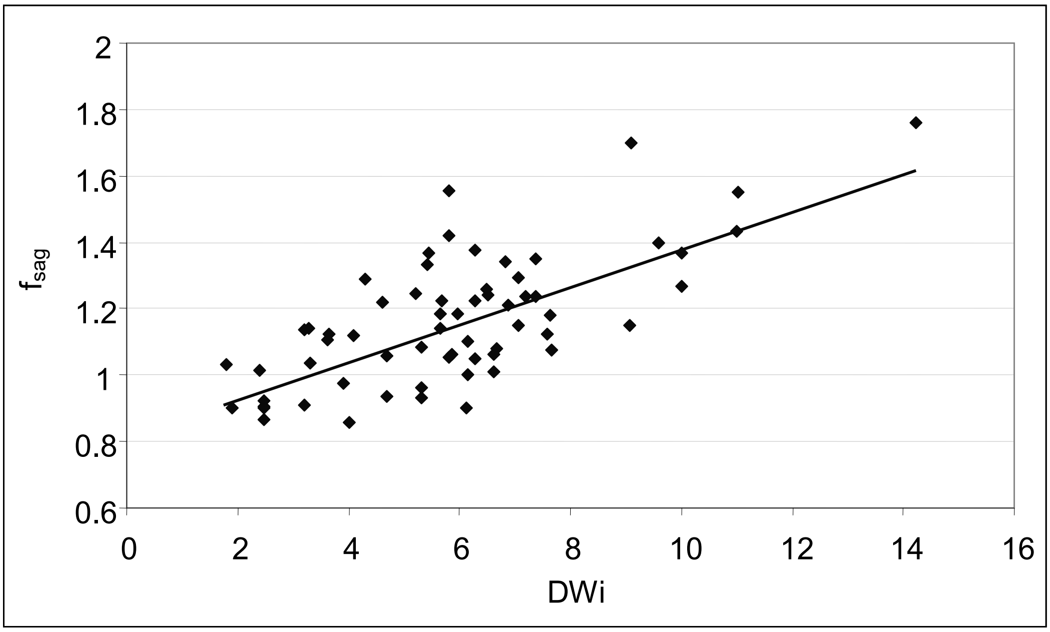


Figure 1 – Measured "fsag" Values vs DWi for a Range of Different Operating Grinding Circuits

Relationships other than Bond's third law have been proposed in the past and include those provided by Kick (1885), Von Rittinger (1867), Charles (1957), Holmes (1957) and Hukki (1962). In this paper one of the more recent relationships (Morrell, 2004[b]) will be used to determine the specific energy of a range of comminution circuits and at the same time assess the relative efficiency of each.

## 2 Equations

### *2.1 General*

In some ways the approach described in the following sections mirrors that of Bond in that it contains a general equation for determining the specific energy to grind rock from a coarser distribution to a finer one as well as work indices related to the strength of the rock. Unlike Bond's approach, where three work indices were defined in relation to particular equipment ie Crushing, Rod milling and Ball milling plus at least 7 "efficiency" factors, the following technique uses only 2 indices related to "coarse" and "fine" ore properties with only one efficiency factor. "Coarse" in this case is defined as spanning the size range from a P80 of 750 microns up to the P80 of the product of the last stage of crushing prior to grinding. "Fine" covers the size range from a P80 of 750 microns down to P80 sizes typically reached by conventional ball milling, ie about 45 microns. The choice of 750 microns as the division between "coarse" and "fine" particle sizes was determined during the development of the technique and was found to give the best overall results across the range of plants in the author's data base. Implicit in the approach is that distributions are parallel and linear in log-log space.

The work index covering grinding of coarse sizes is labelled $M_{ia}$. The work index covering grinding of fine particles is labelled $M_{ib}$. $M_{ia}$ values are provided as a standard output from a SMC Test® (Morrell, 2004[a]) whilst $M_{ib}$ values can be determined using the data generated by a conventional Bond ball mill work index test ($M_{ib}$ is NOT the Bond ball work index). Both of these tests are readily available from mineral processing laboratories around the world.

The general size reduction equation is as follows (Morrell, 2004[b]):

$$W_i = M_i 4\left(x_2^{f(x_2)} - x_1^{f(x_1)}\right) \tag{3}$$

where

$M_i$ = Work index related to the breakage property of an ore (kWh/tonne); for grinding from the product of the final stage of crushing to a P80 of 750 microns (coarse particles) the index is labelled $M_{ia}$ and for size reduction from 750 microns to the final product $P_{80}$ normally reached by conventional ball mills (fine particles) it is labelled $M_{ib}$.

$W_i$ = Specific comminution energy at pinion (kWh/tonne)

$x_2$ = 80% passing size for the product (microns)

$x_1$ = 80% passing size for the feed (microns)

$f(x_j)$ = $-(0.295 + x_j/1000000)$ (Morrell, 2006) (4)

## 2.2 *Specific Energy Determination for Grinding Circuits*

The total specific energy at pinion ($W_T$) to reduce in size crusher product to final product is given by:

$$W_T = W_a + W_b \tag{5}$$

where

$W_a$ = specific energy to grind coarser particles

$W_b$ = specific energy to grind finer particles

Implicit in this approach is the assumption that the grinding specific energy is independent of the processing route and is believed to be applicable to all tumbling mills in the following circuit configurations: crush-rod-ball, crush-ball, crush-HPGR-ball, and AB/SAB, ABC/SABC and single stage AG/SAG circuits.

For coarse particle grinding equation 3 is written as:

$$W_a = KM_{ia}4\left(x_2^{f(x_2)} - x_1^{f(x_1)}\right) \quad (6)$$

where

K = 1.0 for all circuits that do not contain a recycle pebble crusher and 0.95 where circuits do have a pebble crusher

$x_1$ = $P_{80}$ in microns of the product of the last stage of crushing before grinding

$x_2$ = 750 microns

$M_{ia}$ = Coarse ore work index and is provided directly by SMC Test®

For fine particle grinding equation 3 is written as:

$$W_b = M_{ib}4\left(x_3^{f(x_3)} - x_2^{f(x_2)}\right) \quad (7)$$

where

$x_2$ = 750 microns

$x_3$ = $P_{80}$ of final grind in microns

$M_{ib}$ = Provided by data from the standard Bond ball work index test using the following equation (Morrell, 2006):

$$M_{ib} = \frac{18.18}{P_1^{0.295}(Gbp)\left(p_{80}^{\ f(p_{80})} - f_{80}^{\ f(f_{80})}\right)} \tag{8}$$

where

$M_{ib}$ = fine ore work index (kWh/tonne)

$P_1$ = closing screen size in microns

Gbp = net grams of screen undersize per mill revolution

$p_{80}$ = 80% passing size of the product in microns

$f_{80}$ = 80% passing size of the feed in microns

Note that the Bond ball work index test should be carried out with a closing screen size which gives a final product P80 similar to that intended for the full scale circuit.

## 3 Validation

The approach described in the previous section was applied to 65 industrial data sets, a summary of the data base details being given in Table 1. The results are shown in Figure 2. In all cases the specific energy relates to the tumbling mills contributing to size reduction from the product of the final stage of crushing to the final grind. Data are presented in terms of equivalent specific energy at the pinion. In determining what these values were on each of the plants in the data base it was assumed that power at the pinion was 93.5% of the measured gross (motor input) power, this figure being typical of what is normally accepted as being reasonable to represent losses across the motor and gearbox.

Table 1 – Data Base Details

| Circuit | No. | DWi ($kWh/m^3$) | BWib (kWh/t) | F80 (mm) |
|---|---|---|---|---|
| ABC (AG-Ball with pebble crusher) | 2 | 6.3-6.9 | 15.9-16.5 | 85-106 |
| AB (AG-Ball) | 2 | 6.0-6.2 | 9.6-12.1 | 129-134 |
| SS AG (single stage AG) | 6 | 3.3-7.1 | 13.2-19.0 | 100-178 |
| SABC (SAG-Ball with pebble crusher) | 22 | 1.9-11.0 | 10.5-25.0 | 37-176 |
| SAB (SAG-Ball) | 14 | 1.7-14.2 | 9.1-22.9 | 20-212 |
| SS SAG (single stage SAG) | 14 | 1.8-7.1 | 14.0-20.6 | 30-140 |
| Crusher-ball | 3 | 2.5-7.6 | 10.3-18.8 | 8-11 |
| Crusher-rod-ball | 2 | 3.6-7.6 | 10.0-12.4 | 25-35 |

Analysis of the relative error of the predicted specific energy vs the observed values indicated a precision (1 standard deviation) of 7.0%, giving 95% and 99% confidence intervals of approximately +/- 15% and +/- 20% about the mean respectively.

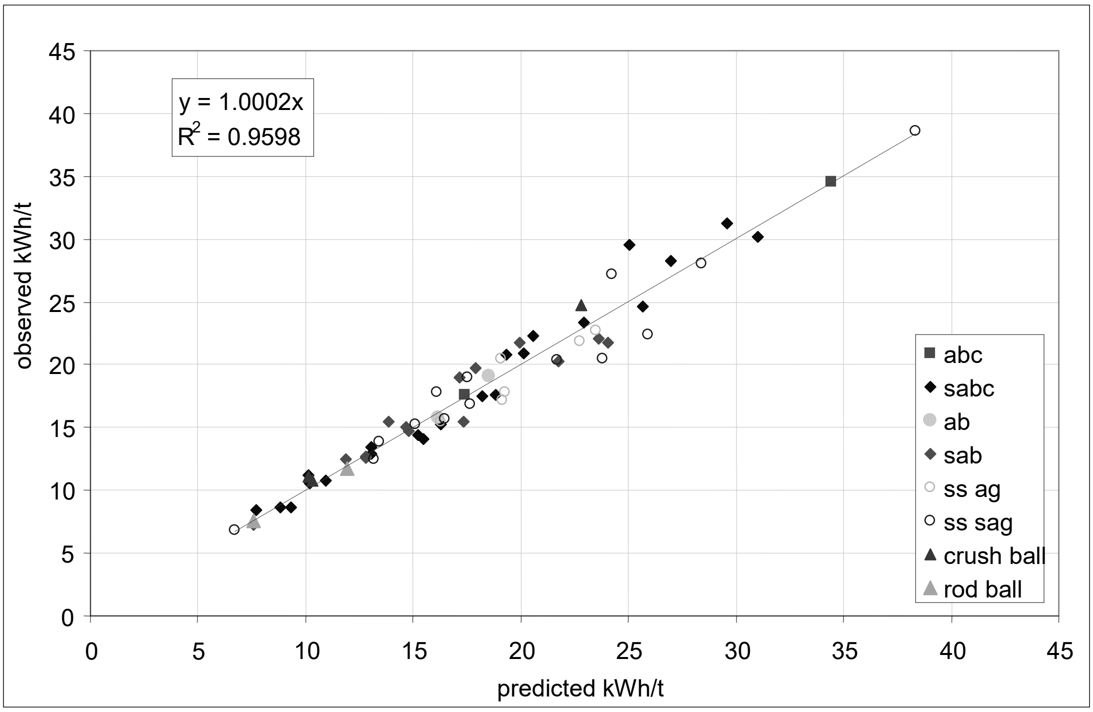


Figure 2 – Observed vs Predicted Total Specific Energy

## 4 Limitations

It is emphasised that $W_a$ and $W_b$ are not the specific energies of individual stages in a grinding circuit, eg the primary and secondary ball mill specific energies respectively or the SAG mill and ball mill specific energies respectively. For example in an AB or SAB circuit $W_a$ is not necessarily the specific energy of the AG/SAG mill circuit and $W_b$ is not the specific energy of the ball mill circuit. In the case of the AG/SAG mill, its specific energy will be a function of ore hardness as well as such factors as feed size, ball charge, aspect ratio, speed and whether it has a pebble recycle crusher or not; its value may be very different to $W_a$. Depending on the combinations of these factors a particular specific energy will result, as will a consequent product size distribution from the AG/SAG mill circuit. Other equations/techniques will need to be applied to determine these values.

The ball mill circuit specific energy will be dictated by the AG/SAG mill circuit product size (so-called transfer size), the final grind size and ore hardness and its value may also be very different to $W_b$. However, it is expected that that the sum of the AG/SAG mill circuit specific energy and the ball mill circuit specific energy will be equal to $W_T$.

It follows from the previous two paragraphs that under no circumstances should it be automatically assumed that the AG/SAG mill transfer size will be 750 microns as a justification for making $W_a$ and $W_b$ the AG/SAG and ball mill specific energies respectively.

The overall specific energy of the tumbling mills only are predicted by this approach. The specific energy of any other size reduction machines in the circuit such as crushers, High Pressure Grinding Rolls (HPGR) or stirred mills have to be determined by other means (Morrell 2006). The total specific energy of the circuit is then found from the sum of the crushing/HPGR energy and that for the tumbling mills plus any stirred mills.

## 5 Energy Utilisation Efficiencies of Different Circuit Designs

### *5.1 General*

The equations used to determine the overall grinding circuit specific energy contain only one circuit-specific factor, that being related to whether there is a pebble recycle crusher in use. This factor has a relatively small influence overall, but never-the-less was indicated to be necessary from comparison of the performances of the circuits with and without pebble crushers. The result is believed to reflect the fact that crushers are more energy-efficient size reduction machines than tumbling mills (Shonert, 1988). In the case of grinding circuits with pebble crushers, the amount of material crushed is typically quite small in relation to new feed and hence the pebble crusher has a relatively small influence on overall energy efficiency. This should not be confused with the often quite significant influence that pebble crushing has on AG/SAG mill throughput (and hence specific energy). The higher throughput that results from using pebble crushing occurs at the expense of the AG/SAG mill circuit grind size which always becomes coarser. This in turn puts more load on the ball mill circuit, which results in the ball mill circuit requiring a higher specific energy to process the extra throughput and maintain grind. The net result is that overall the specific energy may only reduce by a very small amount when the pebble crusher is in use, providing the ball mill circuit is operated to keep the final grind constant. A similar response is also seen with the introduction of pre-crushing in SAG mill circuits ie it results in higher SAG mill throughputs but with a coarser grind.

Apart from pebble crushing, any other differences in energy utilisation efficiency inherent in particular grinding circuit configurations and/or machine types should be seen as systematic trends in the relative errors of the equations. In Figure 2 this would show up as data points from particular circuits tending to be above or below the line of equality. No such trends were found and hence there appears to be no bias with respect to the type of grinding circuit within the data base.

### *5.2 Influence of Crushing/HPGR*

The overall comminution energy of a circuit is obtained by summing the specific energy of the grinding mills plus that associated with any crushers and HPGRs in the circuit. The question then arises: when the crushing energy is taken into consideration are certain circuit designs found to have an overall higher energy efficiency? On the basis of the author's data base the answer appears to be that apart from the previously mentioned minor influence of pebble crushing there is no evidence to statistically prove that any circuit design is consistently superior from an energy efficiency viewpoint. From the literature, individual cases can be identified where the introduction of a particular circuit design appears to have resulted in a drop in the overall circuit specific energy, eg at ASARCO's Ray concentrator (McGhee et al, 2006) and at St Ives (Atasoy et al, 2006) the authors claimed that incorporating pre-crushing ahead of milling improved energy efficiency. This may be true, but inspection of their data indicates that the final grind of the circuits when pre-crushing was used was much coarser than when pre-crushing was not utilised, making conclusive comparisons of energy efficiency almost impossible. This is in contrast with the experiences at the Haveluck/Paddy's Flat complex (Morrell et al, 1991) and at Brunswick (Larsen et al, 2001) where it was concluded that multi-stage crushing followed by rod and/or ball milling gave the same overall specific energy as AG and SAG mill circuits.

Given that it is believed that crushers and HPGRs are more energy efficient than tumbling mills (Schonert, 1988) and that there is evidence in the author's data base that pebble crushing improves energy efficiency (albeit by a small amount) it should follow that circuits with a higher proportion of energy used via crushing should be more energy efficient. A good example of this is the use of HPGRs which, from pilot testing at least, appear to result in an overall drop in energy usage when compared to AG and SAG milling (Parker et al, 2001). The apparent absence of conclusive evidence concerning crushing circuits in the author's data base may be due to the fact that a significant reduction in overall energy usage will only be observed where the proportion of total energy used in crushing becomes relatively high. Where the relative amount of crushing energy is small only small differences in overall efficiency may result. Hence in such cases observable trends will only be apparent with a very large base of high quality relevant data. The amount of energy used for crushing only becomes relatively large when a relatively fine crushed product is produced, as is the case with HPGR circuits, and/or the ore is hard from a crushing perspective. Hence the influence on energy efficiency is more likely to be readily apparent in such cases. The DWi is highly correlated with UCS and the point-load index, which are traditional indicators of hardness from a crushing perspective. Where the DWi is high, therefore, it might be expected that crushing circuits (including HPGR circuits) would show measurable energy efficiency gains compared to AG/SAG circuits. This is effectively what Figure 1 implies and the HPGR testwork reported by Parker et al (2001) for Boddington Gold demonstrates. In this case the DWi of the ore was in the range 9-13 kWh/$m^3$ and the crushing/HPGR part of the circuit accounted for over 17.5% of total power and hence provided a large contribution to the overall comminution energy demand. In contrast the ore treated by the Paddy's Flat and Haveluck circuits (Morrell et al, 1991) had a relatively low DWi value of 2.5 kWh/$m^3$ and in the crushing-ball mill plant (Haveluck) the crushing circuit only accounted for 7% of total comminution energy. The overall specific energy of this plant was therefore found to be very similar to the SABC configuration of Paddy's Flat.

### *5.3 Problems with Interpreting Comparative Data*

Unless different circuit designs are run under optimal conditions using the same feed ore, feed size distribution and grinding to the same final product size distribution it is almost impossible to draw any firm conclusions concerning their relative energy efficiencies. This is typified by the studies reported by Levin (1992) and Mainza and Powell (2006). In his study, Levin collated the results from an unpublished survey carried out by Mintek in 1988 of a number of different comminution circuits treating Witwatersrand gold ores. He grouped together the results from 43 of the plants studied into 3 circuit designs:

Circuit 1 - primary rod mill followed by secondary pebble milling
Circuit 2 - primary ball mill followed by secondary pebble milling
Circuit 3 – single stage SAG milling

He made the assumption that the average Bond operating work indices from each circuit should indicate their relative energy utilisation efficiencies as he maintained that the ore breakage characteristics of all Witwatersrand gold ores is very similar (an assumption in the author's experience which is believed to be unlikely to occur in practice). The resulting Bond operating work indices reported in his paper are as given in Table 2. As can be seen the Bond operating work index for the rod-pebble mill circuit was found to be the lowest and that for the SAG mill circuit was the highest, suggesting that the SAG mill circuit is the least energy efficient. Also shown in the same table are the operating work indices using equation 3 as the basis for calculation ($MOW_i$ values). These values suggest that the ball-pebble mill circuit is the worst from an energy efficiency viewpoint and that the rod-pebble mill circuit is only very marginally better than the SAG mill. In fact the $MOW_i$ value differences are quite small and statistical comparison concludes that there are no apparent differences between the circuits.

Table 2 – Comparison of the Efficiencies of 3 different circuits (data after Levin, 1992)

| | circuit 1 | circuit 2 | circuit 3 |
|---|---|---|---|
| F80 (microns) | 12700 | 10200 | 111000 |
| P80 (microns) | 89 | 87 | 90 |
| BOWi (kWh/t) | 19.2 | 20.3 | 22.4 |
| MOWi (kWh/t) | 22.4 | 24.3 | 23.0 |

Mainza and Powell's data relate to a comparison of a mill circuit run in closed-circuit autogenous mode and in so-called Run-of-Mine ball mill mode, which is effectively single stage ball milling with a relatively coarse feed. The original intention of the study appears to have been to compare the performance of the mill under these two modes of operation from a modelling perspective and to this end attempts were made to run the mill at the same/similar feed rate. Their data are reproduced in Table 3 and regardless of whether the circuits are compared using $BOW_i$ or $MOW_i$ values there is clearly a large apparent difference in efficiency, autogenous operation appearing to be vastly superior to ball milling. The problem with this study is that the RoM ball mill circuit appears to have been grossly underfed. Mainza and Powell reported that the mill had a 30% ball charge, resulting in a power draw of almost 10 MW compared with when the mill was operated in autogenous mode where it drew 5 MW. However, the feedrate in RoM ball mill mode was only increased by 10% on average from the autogenous mode condition. Ball mills and SAG mills need to be operated with a sufficient rock charge such that ball-on-ball collisions are minimised as otherwise this results in energy being wasted on grinding the steel media rather than the feed ore. Mainza and Powell's reported data indicate that the rock charge was extremely small in one survey and non-existent in the other. Hence much of the mill power may have been wasted on ball-on-ball contacts. Further indications of inefficient operation in the RoM ball mill mode was that the mill discharge P80 and the classifier undersize were almost identical, suggesting that the recycle load was very small. It is believed that throughput could have been greatly increased in RoM ball mill mode (as a result the rock load would have been much higher) and quite possibly brought up to a level where the operating work index would have more closely matched that from autogenous operation. Some evidence for this exists within Mainza and Powell's data as in the 2 surveys that they carried out of the RoM ball mill circuit one was done at a 10% higher through than the other which resulted in a Bond operating work index which was 10% lower.

Table 3 - Comparison of the Efficiencies of 2 different circuits (data after Mainza and Powell, 2006)

| | AG | RoM Ball |
|---|---|---|
| Tph | 679 | 756 |
| Power (kW) | 5015 | 9925 |
| F80 (microns) | 43850 | 33350 |
| P80 (microns) | 314 | 216 |
| BOWi (kWh/t) | 14.3 | 21.0 |
| MOWi (kWh/t) | 11.8 | 19.1 |

## 6 Influences of Operating Conditions on Energy Utilisation Efficiency

In closed ball mill circuits, underfeeding generally results in relatively low circulating loads and conversely a circuit which is being operated at its maximum throughput potential will tend to have a relatively high recycle load. Avoiding a low recirculating load is therefore generally considered as being desirable as higher energy utilisation efficiencies are thought to result with increased circulating load. However, as Hukki pointed out (1980), higher recirculating loads must be obtained through higher feedrates and with no deterioration in classifier performance. Collecting data from operating plants to test these views is notoriously difficult. Experiments were therefore carried out using the Bond laboratory ball work index procedure. In these experiments the same material was tested using recirculating loads of 150%, 250% and 400%. As the material was the same in each case, any differences in the work indices that result would be due to changes in the manner that the tests were done, ie changing recirculating load. The resultant work indices are shown in Figure 3 and are represented relative to the lowest work index obtained. As can be seen, as the recirculating load increased the work index reduced, indicating an apparent steady increase in efficiency. However, analysis of the final product size distributions from each test (Figure 4) shows that the shape of the distribution also changed systematically, the gradient being steeper at the higher recycle loads. The indications from these tests are that the higher recycle loads do not result in greater efficiencies from the viewpoint that more of the energy in the balls is transferred into breaking feed ore but that the same amount of energy is used to break more of the coarser ore particles and less of the finer ones, thereby changing the product size distribution. Therefore the net result of increasing throughput and operating the classifier with the same efficiency but maintaining a constant P80 should be an increase in recycle load and less overgrinding of the finer fractions. The systematic change in the work index is therefore due to a progressive departure from the underlying requirement of the energy-size relationship for the feed and product size distribution to remain parallel in log-log space.

As stated previously increasing recycle load has to be the result of increasing throughput for an increase in apparent efficiency to result. Hence increasing recycle load by causing the classifier performance to deteriorate is not expected to give the same result. To illustrate this, the Bond laboratory ball work index test was once again utilised to conduct a series of tests. Four tests were carried out on the same ore and in each successive test an increasing proportion of mill discharge was recycled to the mill without being classified on the closing screen. This was done to reproduce a range of classifier efficiencies through varying the amount of by-pass. The resultant trend in relative work index is shown in Figure 5. A systematic and significant increase in work index is apparent as the amount of classifier by-pass is increased. As Figure 6 illustrates, the effect on the gradient of the final product size distribution is obvious. As the efficiency of the classifier improves (by-pass reduces) the gradient of the final product size distribution becomes steeper.

Combining the results from these two series of tests and making allowances for the fact that plant conditions never match the ideal environment that laboratory test are undertaken in, it is believed that there is a MAXIMUM potential (ie from very poor practice to best practice) of about 7% increase in apparent energy utilisation efficiency by accommodating a significant increase in recycle load through increasing throughput, and a similar MAXIMUM potential by improving classification efficiency. These effects need to be considered in the light of the results shown in Figure 2 and the application of the approach described in Section 2 as they could account for a large proportion of the observed scatter.

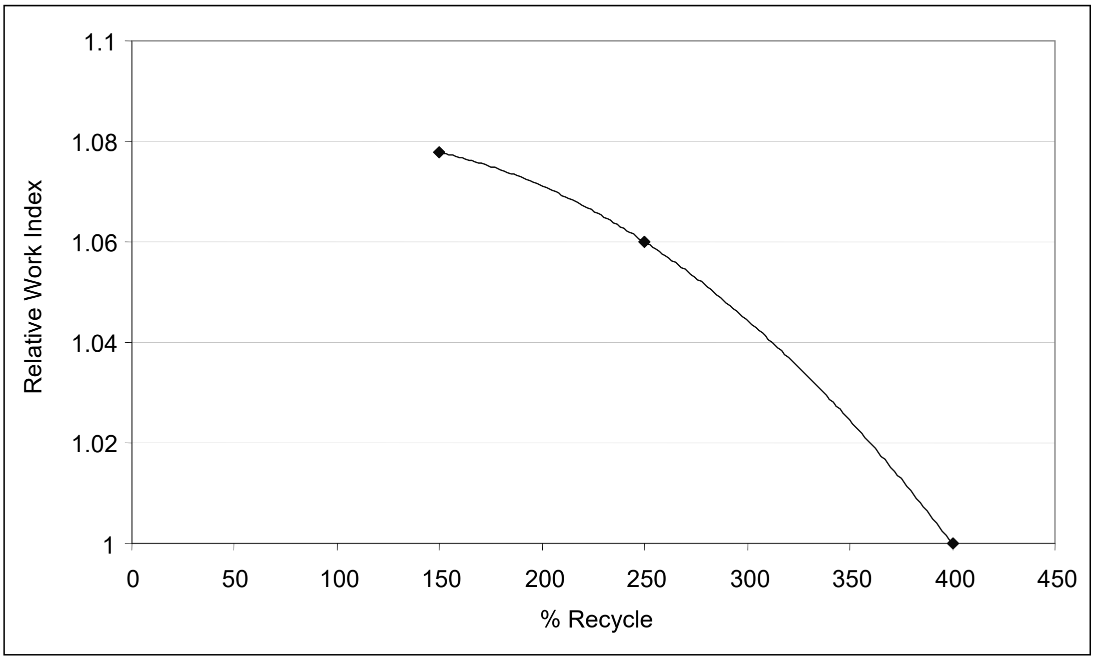


Figure 3 – Relative Work Indices Obtained from Bond Laboratory Work Index Tests Carried Out With Different Recycle Loads

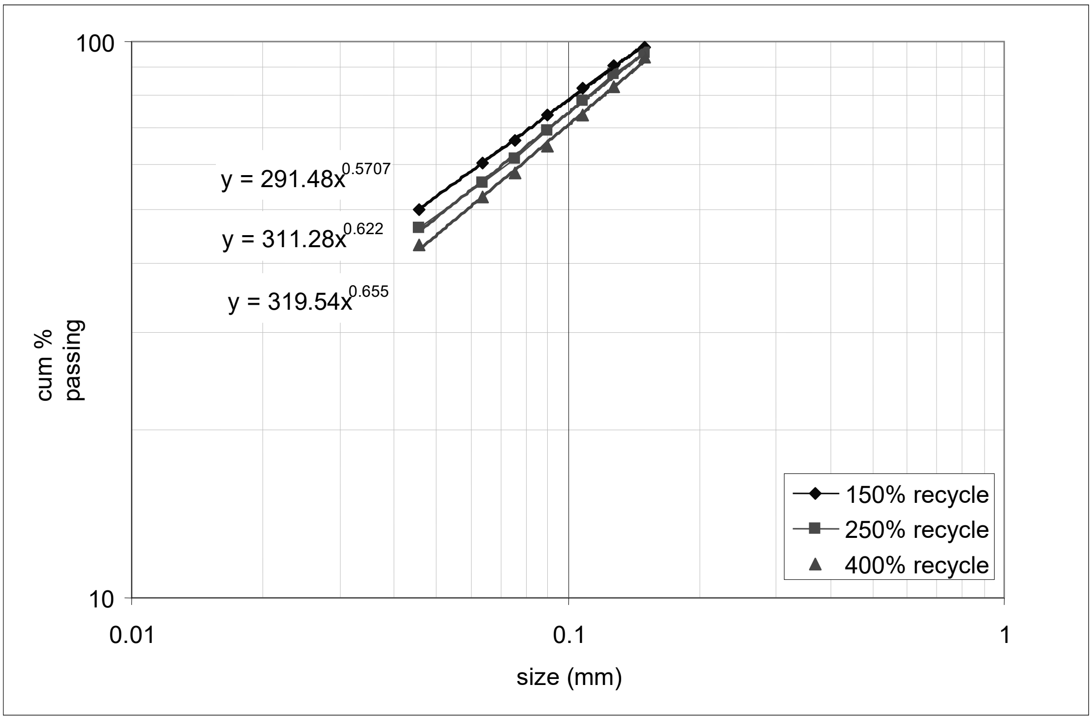


Figure 4 – Final Product Size Distributions Obtained from Bond Laboratory Work Index Tests Carried Out With Different Recycle Loads

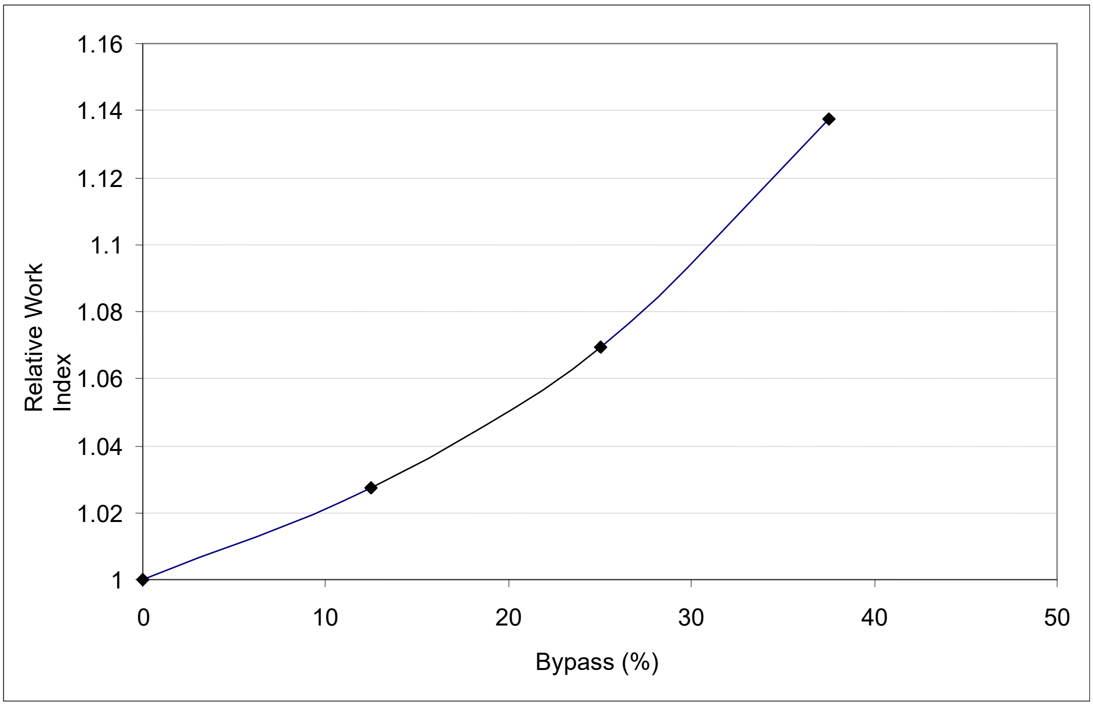


Figure 5 – Relative Work Indices Obtained from Bond Laboratory Work Index Tests Carried Out With Different Classifier Performances

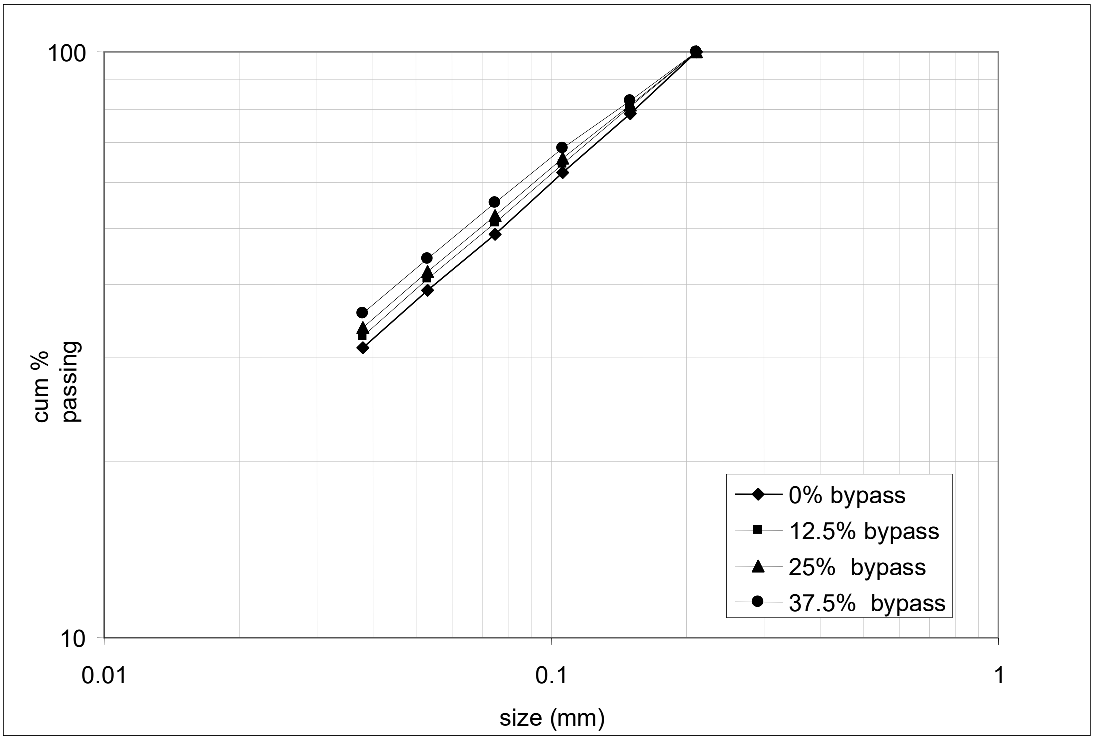


Figure 6 – Final Product Size Distributions Obtained from Bond Laboratory Work Index Tests Carried Out With Different Classifier Efficiencies

# 7 Conclusions

By analysing data from 65 different comminution circuits using relatively simple energy-size reduction equations, it was concluded that within the limits of the precision of the equations used (7.0 %) there were no significant differences between the energy utilisation efficiency of any of the grinding circuits other than that caused by the use of pebble recycle crushers. On average this was found to result in a small, though measurable improvement in energy utilisation efficiency.

These equations use only 2 rock characterisation parameters, one describing coarse rock grinding ($M_{ia}$) and the other fine rock grinding ($M_{ib}$). The former is provided from a SMC Test® whilst the latter is derived from the data supplied by a standard Bond laboratory ball work index test.

Laboratory testwork indicated that the way circuits are operated with respect to classifier performance and recycle load in closed ball mill circuits could account for differences in apparent energy utilisation efficiency of a maximum of 15% from worst to best case conditions. The experimental data indicated that this was possible through changes in the gradient of the final product size distributions.

The author's data base contains a number of multi-stage crushing rod/ball mill circuits. When compared to the performance of AG/SAG mill circuits there was no indication that overall they had and better energy utilisation efficiencies. It is believed that due to the limited number of these circuits in the data base (5) this conclusion should not be generalised. It is further believed that improvements in overall energy efficiency may result with the use of multi-stage crushing or HPGR size reduction in cases where the ore is very competent (high DWi values) and where the crushing and/or HPGR component of total energy consumption is relatively high.

# 8 References


Atasoy, Y., Valery, W. and Skalski, A. 2001. Primary Versus Secondary Crushing at St. Ives (WMC) SAG Mill Circuit. Proc International Autogenous and Semi Autogenous Grinding Technology,Vancouver, vol I pp 248-261

Barratt D J and Allan M J. 1986. Testing for autogenous and semiautogenous grinding: A designer's point of view. Minerals and Metallurgical Processing, May, pp 65-74

Bond, F. C.,1952. The third theory of comminution, Trans AIME, vol 193, pp 484-494

Bond, F.C., 1961. Crushing and Grinding Calculations. Brit. Chem Eng. Part I, 6 (6), 378-385, Part II, 6 (8), 543-548

Charles R J 1957 Energy-size reduction relationships in comminution Trans SME/AIME vol 208 pp 80-88

Holmes J A 1957 A Contribution to the study of comminution – a modified form of Kick's Law Trans Instn Chem Engrs vol 35 pp 126-156

Hukki, R.T., 1962. Proposal for a solomnic settlement between the theories of von Rittinger, Kick and Bond, Trans AIME , vol 223, pp403-408.

Hukki R T., 1980 Fundamentals of the Closed Grinding Circuit E&MJ second operating handbook of mineral processing, edited by Lane White N.Y., E/MJ Mining Information Services ; McGraw Hill, pp 65a-65 h

Kick, F.,1885. Das Gesetz der proportionalen Widerstande und seine anwendung felix, Leipzig

Knight, D A, Medina V G and Babcock G H. 1989. Comminution Circuit Comparison – Conventional vs Semi Autogenous. Proc International Autogenous and Semi Autogenous Grinding Technology,Vancouver, pp217-224

Larsen, C., Cooper, M. and Trusiak, A.,2001. Design and Operation of Brunswick's AG/SAG Circuit, Proc International Autogenous and Semi Autogenous Grinding Technology,Vancouver, vol IV pp 350-367.

Levin J 1992 Indicators of grindability and grinding efficiency, J. S. Afr. Inst. Min. Metall vol 92 no 10 pp 283-289

Mainza A and Powell M S, 2006 RoM Ball mills – a comparison with AG/SAG Milling. Proc International Autogenous and Semi Autogenous Grinding Technology,Vancouver, vol II pp 314-325

McGee, S. Mosher, J., Richardson, M., David, D. and Morrison, R., 2001. SAG Feed Pre-Crushing at ASARCO's Ray Concenetrator: Development, Implementation and Evaluation. Proc International Autogenous and Semi Autogenous Grinding Technology,Vancouver, vol I pp 234-247

Morrell, S., 2006. Rock Characterisation for High Pressure Grinding Rolls Circuit Design, Proc International Autogenous and Semi Autogenous Grinding Technology,Vancouver, vol IV pp 267-278.

Morrell, S., 2004[a]. Predicting the Specific Energy of Autogenous and Semi-autogenous Mills from Small Diameter Drill Core Samples. Minerals Engineering, Vol 17/3 pp 447-451

Morrell, S., 2004[b]. An Alternative Energy-Size Relationship To That Proposed By Bond For The Design and Optimisation Of Grinding Circuits. International Journal of Mineral Processing, 74, 133-141.

Morrell, S., Johnson, G. and Revy, T., 1991. A comparison through observation and simulation of the power utilisation and performance of two dissimilar comminution plants. Fourth Mill Operators' Conference, Aus.IMM Burnie, pp157-160. March

von Rittinger, P. R., 1867. Lehrbuch der Aufbereitungskunde, Ernst and Korn, Berlin

Thomas K G., 1989. SAG Mills and Barrick Gold. Proc International Autogenous and Semi Autogenous Grinding Technology,Vancouver, pp229-251

Parker, B., Rowe, P., Lane, G. and Morrell, S., 2001. The Decision to Opt for High Pressure Grinding Rolls for the Boddington Expansion. Proc International Autogenous and Semi Autogenous Grinding Technology,Vancouver, vol III pp 93-106

Schonert, K., 1988. A first survey of grinding with high-compression roller mills. International Journal of Mineral Processing, 22(1-4): 401-412.

**Appendix 1 – Worked Example**

A SMC Test® and Bond ball work index test were carried out on a representative ore sample. The overall grinding specific energy to reduce a primary crusher product with a P80 of 100 mm to a final product P80 of 75 microns is required to be estimated. No crushing of recycle pebbles is to be included in the circuit.

*A.1 Determination of Work Indices*

Coarse particle grinding work index ($M_{ia}$):

$M_{ia}$ = 19.5 kWh/t (provided by SMC Test®)

Fine particle grinding work index ($M_{ib}$):

From the Bond ball work index test results the following values were obtained:

Gbps = 1.3 gms/rev

$f_{80}$ = 2250 microns

$p_{80}$ = 78 microns

$P_1$ = 106 microns

Note that the Bond test was correctly done to produce a final product similar to that required of the full scale circuit.

From eq 4:

$f(p_{80})$ = -(0.295+78/1000000)

= -0.2951

$f(f_{80})$ = -(0.295+2250/1000000)

= -0.2973

From eq 8

$$M_{ib} = \frac{18.18}{106^{0.295}(1.3)\left(78^{-0.2951} - 2250^{-0.2973}\right)}$$

= 20.1 kWh/t

## *A.2 Determination of Grinding Circuit Specific Energy*

Primary Crusher Product $P_{80}$ = 100mm

Final Product $P_{80}$ = 75 microns

<u>Coarse particle grinding specific energy</u>

Combining eq 4 and 6:

$$W_a = 1*19.5*4\left(750^{-(0.295+750/1000000)} - 100000^{-(0.295+100000/1000000)}\right)$$

$$= 10.2 \text{ kWh/t}$$

<u>Fine particle grinding specific energy</u>

Combining eq 4 and 7:

$$W_b = 20.1*4\left(75^{-(0.295+75/1000000)} - 750^{-(0.295+750/1000000)}\right)$$

$$= 11.1 \text{ kWh/t}$$

<u>Total grinding specific energy at pinion</u>

From eq 5:

$W_T$ = 10.2 + 11.1 kWh/t

= 21.3 kWh/t

The 95% confidence limits are 18.1 - 24.5 kWh/t

If the measured plant grinding specific energy (after power @ pinion allowance) is outside of these limits it is possible that:

1. There are operational problems with the plant causing relatively large inefficiencies, eg poor classification or underfeeding of the circuit.
2. Problems with inaccurate power measurements or that the measured power as indicated at the DCS is not motor input power but has already been adjusted to reflect power at pinion.
3. Problems with ore sample selection and/or subsequent laboratory testing.
4. Problems with measurement of the final grind P80.

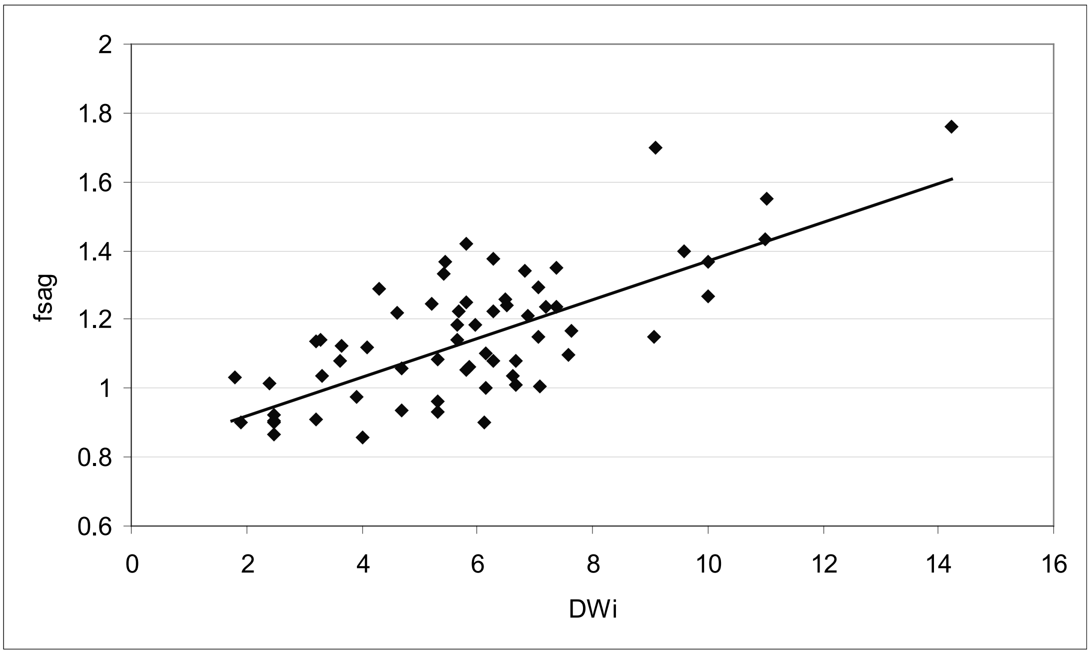


Fig 1

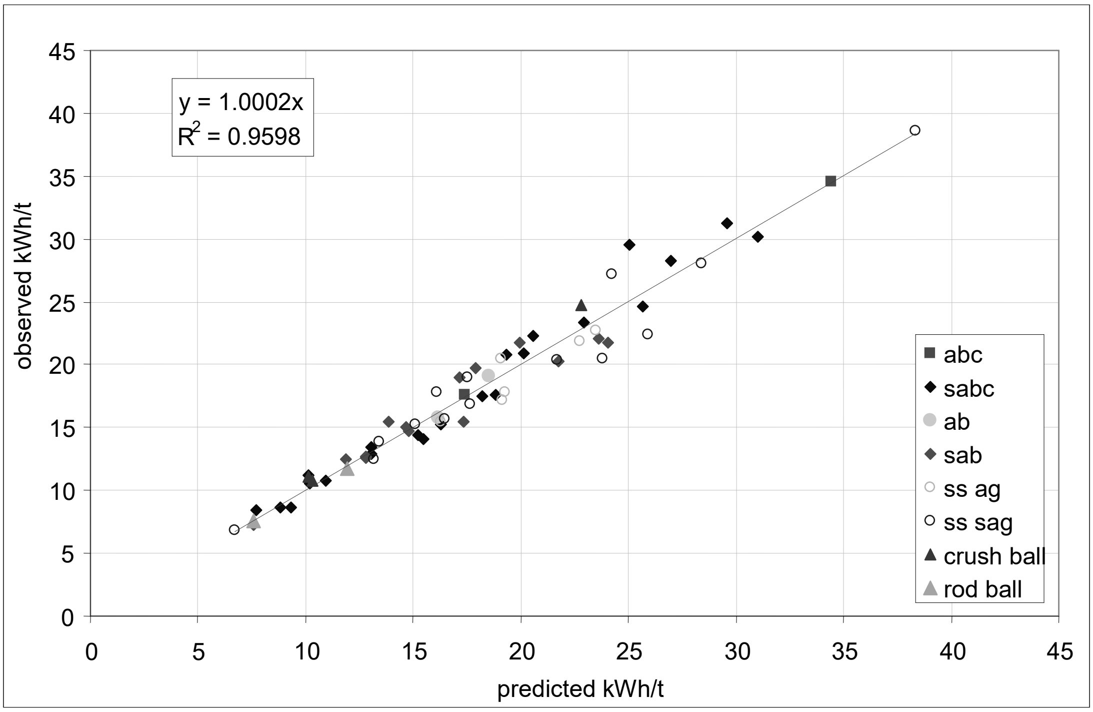


Figure 2

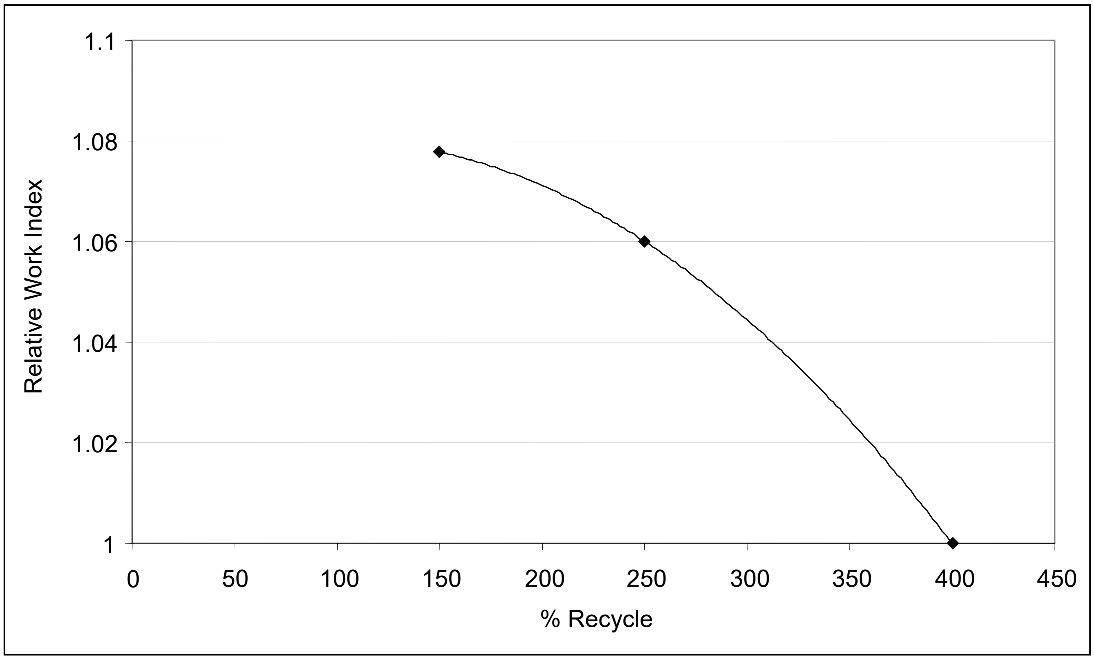


Figure 3

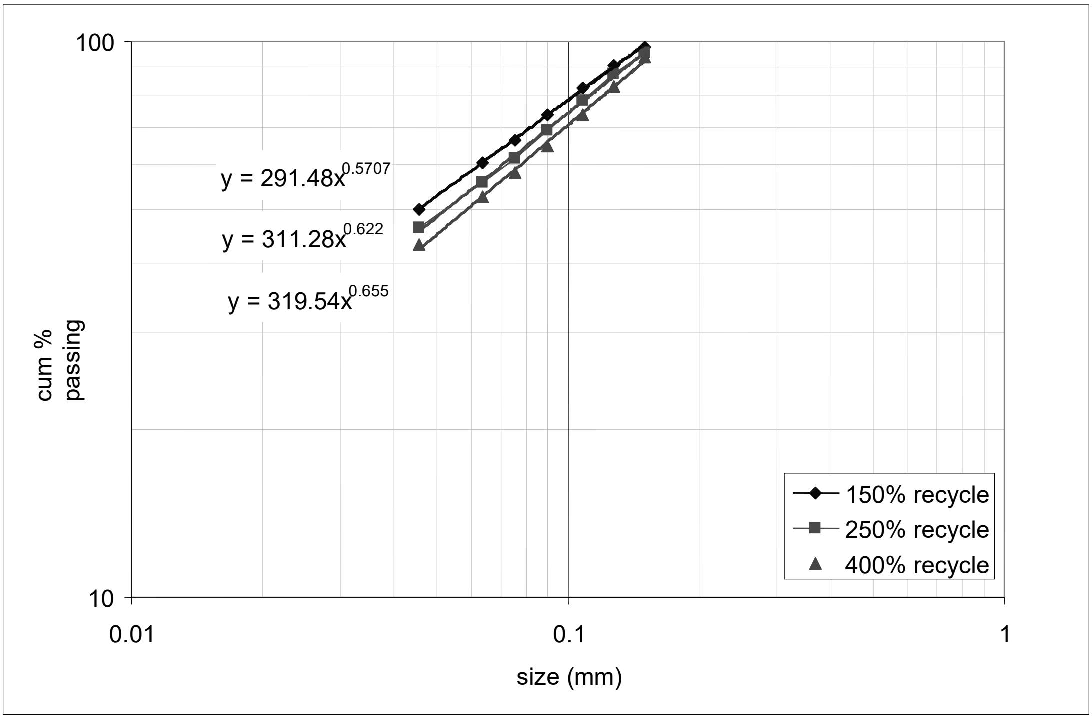


Figure 4

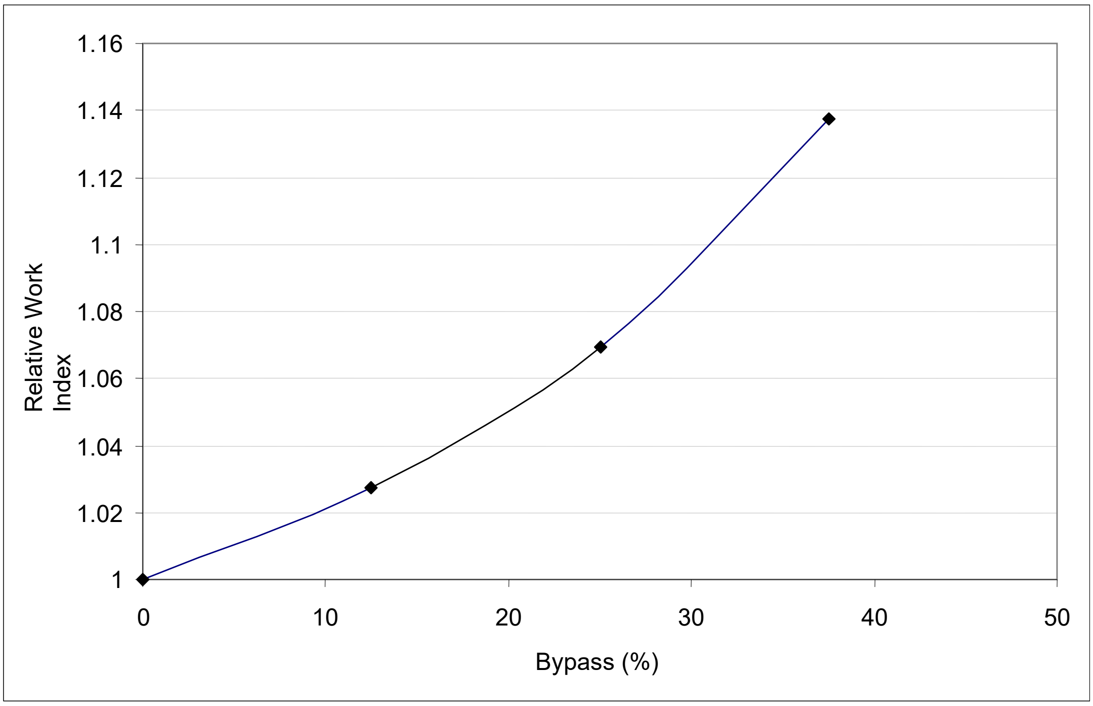


Figure 5

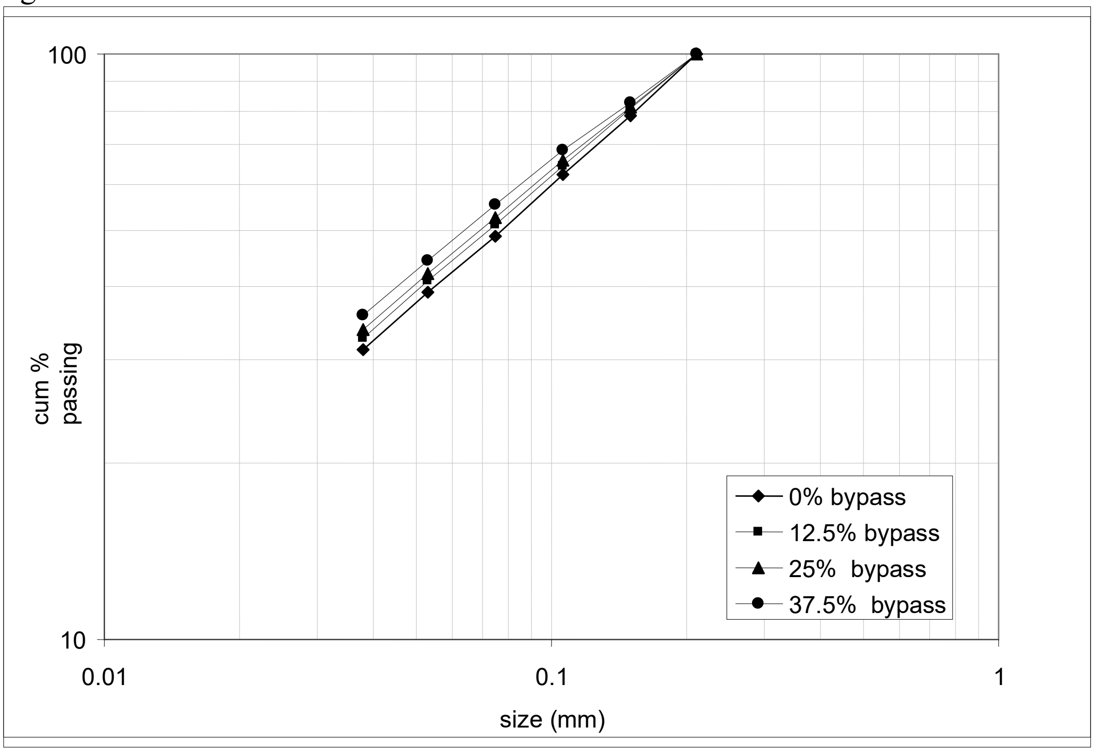


Figure 6